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MINING PRACTICE AND COSTS AT THE VIPOND MINE, TIMMINS, ONTARIO, CANADA

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BY

ROBERT E. DYE

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MINING PRACTICE AND COSTS AT THE VIPOND MINE, TIMMINS, ONTARIO, CANADA[1]

By Robert E. Dye[2]

INTRODUCTION

This paper, describing the mining methods employed and setting forth prevailing costs at the property, is one of a series being prepared by the United States Bureau of Mines dealing with properties in various mining districts of the United States and Canada.

The Vipond mine is a gold property located at Timmins, Ontario, Canada, and operated by the Vipond Consolidated Mines (Ltd.). For the past fiscal year (ended July 31, 1930) ore was mined and milled at the rate of 310 tons per day, and the value of the bullion produced was $896,397.61.

ACKNOWLEDGMENTS

The author acknowledges the assistance rendered by the mine staff in compiling data for this paper. He is indebted also to Charles F. Jackson, principal mining engineer, U. S. Bureau of Mines, for many valuable suggestions.

HISTORY AND LOCATION

The Vipond mine is located in the Porcupine gold area of northern Ontario, 1 mile south of the town of Timmins. The district, which is approximately 500 miles north of Toronto, in the Hudson Bay drainage basin, is served by the Temiskaming and Northern Ontario Railway. The area may also be reached, except in the winter months, by auto over the Ferguson highway.

Early geological reports[3] mention the occurrence of gold in small quantities, but it was not until 1909 that discoveries were made which led to the immediate opening up of the area.

In the following year, 1910, the initial gold production from the camp amounting to $35,539, was recorded. Since that time much progress has been made. For the year ended December 31, 1929, the total production from the district amounted to $19,281,286. The total production from the camp to the end of 1929 has amounted to $247,392,353.

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2 One of the consulting engineers, U S Bureau of Mines, and general superintendent of Vipond Consolidated Mines (Ltd)

3 Burwash, Edward M , Geology of the Nipissing-Algoma Line: Bull 6, Ontario Bureau of Mines 1896 pp 167-184
Parks, William A , Report on Niven's Base Line: Bull 9, Ontario Bureau of Mines, 1900. pp 125-142

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GEOLOGY

The geology of the field has been described in detail by A. G. Burrows,[4] now provincial geologist. The region is located in the midst of a huge peneplane over which passed the Pleistocene glaciers. The local topography is not rugged, the highest hills rising less than 200 feet above the surrounding country. The lesser valleys are well marked, though in their lower reaches they are often muskeg and swamp.

Outcrops are fairly numerous. The overburden, which is usually not heavy, is glacial detrital material consisting of bowlders, sand, gravel, and clay.

The rocks are of pre-Cambrian age, the important rocks locally being a complex extrusive series consisting of basic to acid lava flows which have been extensively and intensively folded and distorted, and later intruded by quartz porphyry. These movements have resulted in extensive shearing and widespread schistosity of the rocks. Ore deposits have been localized in the resulting zones of structural weakness, in part along the margins of the porphyry and in part in various lava beds which have proved to be structurally favorable, due largely, if not entirely, to their physical properties.

The Vipond mine is located at the nose and northern leg of a synclinal fold, the axis of which strikes a few degrees north of east, and plunges toward the east at about 55° from the horizontal. The mine, located as it is in an unusually contorted and distorted area, yields orebodies which are erratic as to occurrence and have a wide variation both as to their attitude and form. This feature will become apparent by comparing Figure 1, which shows the orebodies and mine workings at the 400-foot level, with Figure 2, which shows the corresponding features in the same zone at the 600-foot horizon.

NATURE OF THE ORE

The ore consists of quartz veins and irregular masses of mineralized schist impregnated with quartz. A great variety of sizes and shapes of ore masses are encountered and mined. Variations from the narrowest workable widths up to as much as 60 feet have been dealt with. Lengths vary from 10 or 12 feet up to 400 feet. The dips, generally speaking, are more than 60° from the horizontal, though a few flat veins, standing at about 30° from the horizontal, have been mined. The erratic nature of the ore occurrences makes it vital to investigate by raise or drift every exposure which shows any promise.

The ore is quite hard and stands well. The walls of the deposits are generally well defined but in part are "assay walls." The enclosing rocks, except where they are badly sheared or consist of the graphitic material with which much of the ore is associated, are firm and stand well.

The ore contains 10 to 30 per cent quartz and 5 to 8 per cent of sulphides. Of the sulphides, by far the greater proportion consists of iron pyrite with which is associated small amounts of sphalerite, galena, and chalcopyrite. Only a very small proportion of the gold is visible, by far the greater proportion being very finely disseminated throughout the mass and favoring the pyrite, the quartz, and the schist in the order named.

4 Burrows, A. G., The Porcupine Gold Area: Ontario Department of Mines, vol. 33, part 2, 1924, 105 pp.

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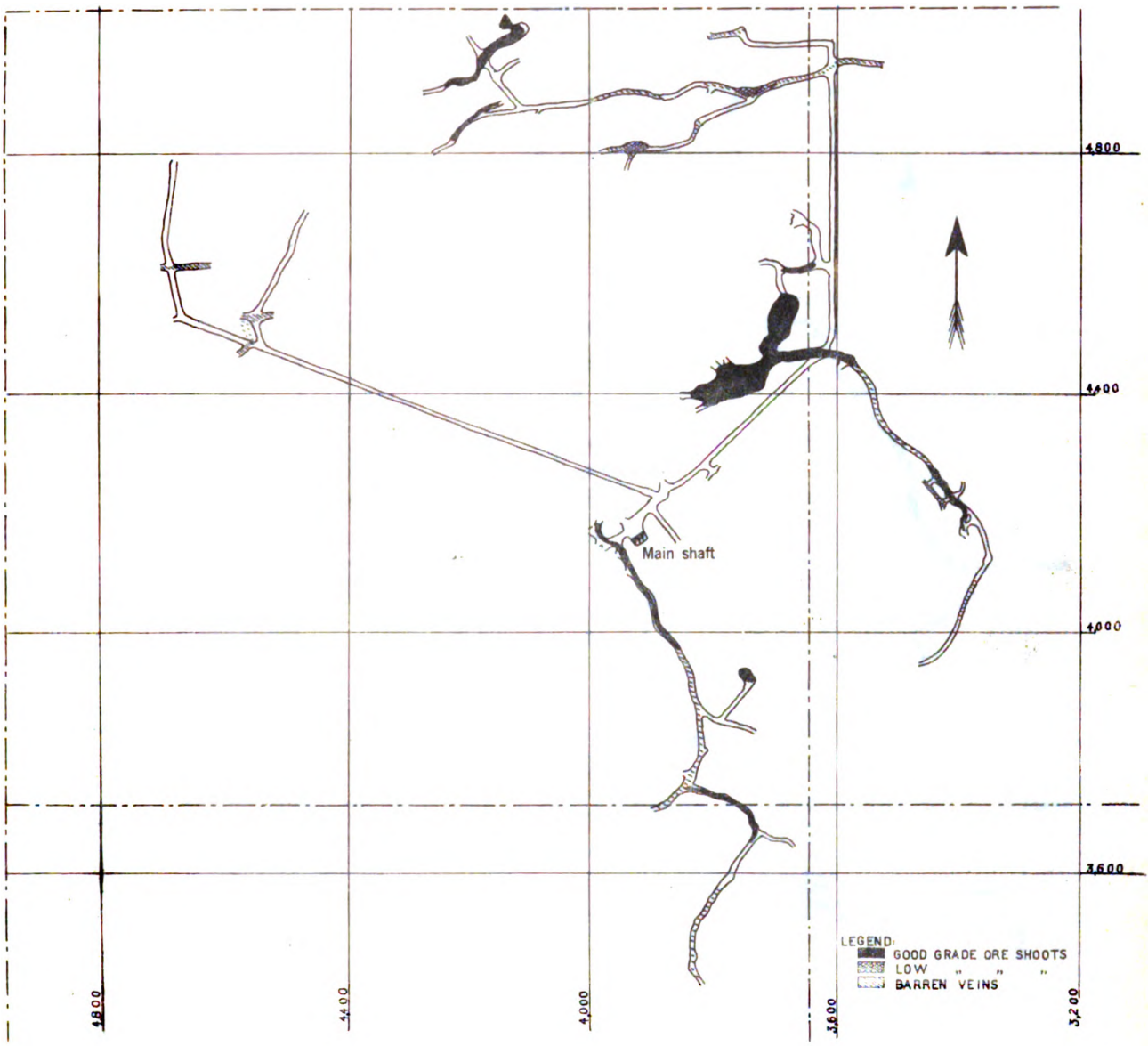

Figure 1.—Orebodies and mine workings, 400-foot level

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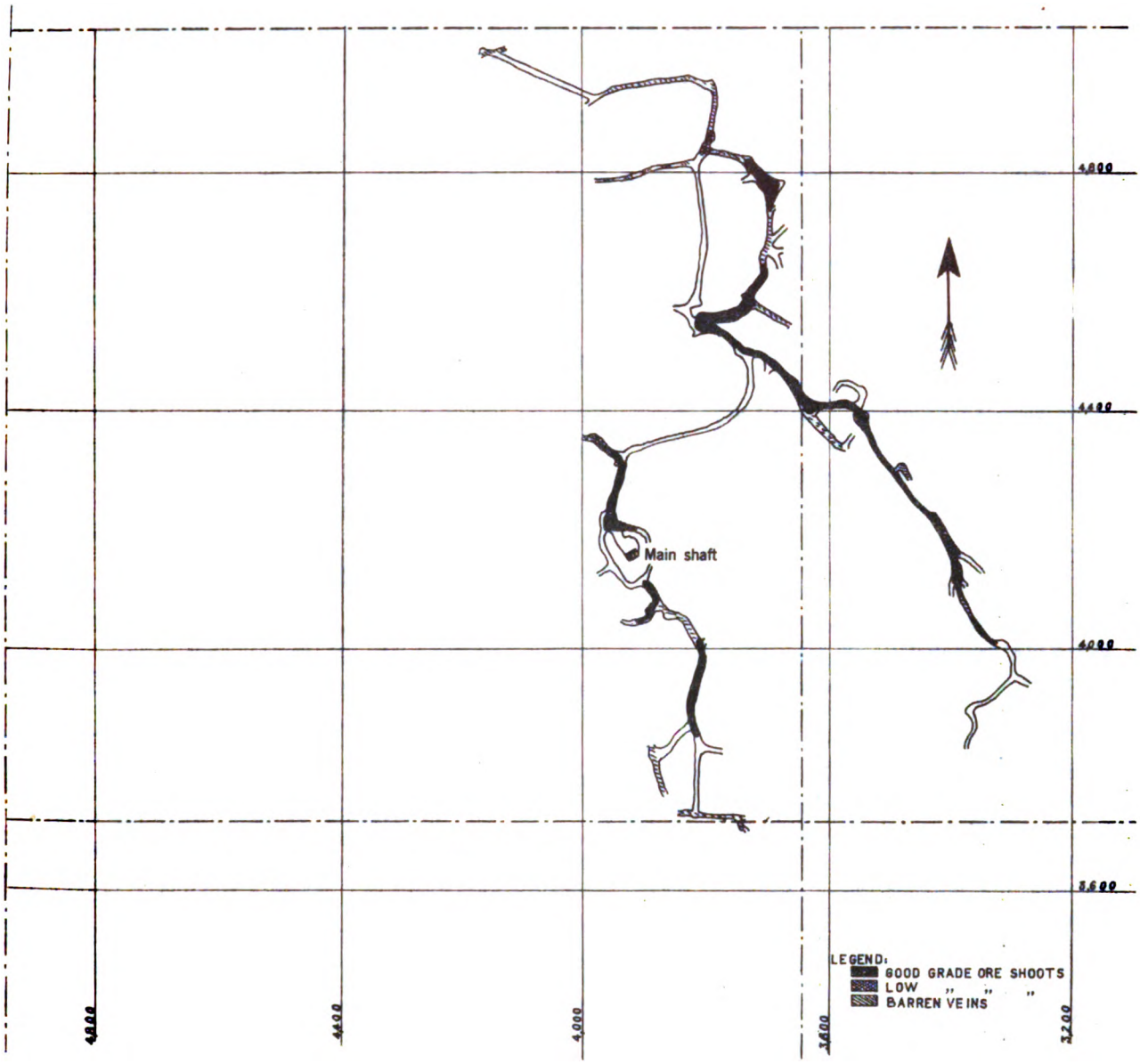

Figure 2.—Orebodies and mine workings, 600-foot level

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METHODS OF PROSPECTING AND EXPLORATION

Early prospecting was done by surface trenching, pop shots, shallow pits, an diamond drilling from the surface. Later and more productive exploration has been accomplished by work done from below ground, as by far the greater number of the payable veins apex below the surface. A great deal of diamond drilling, mostly horizontal holes, is done below ground from drifts and crosscuts to serve as a guide for subsequent excavation work. Much valuable information as to structure and possible ore occurrence is obtained from this drilling, though little dependence can be placed upon results as to the probable importance of the ore indications encountered. The drilling serves, at a relatively small cost, tc eliminate areas which would otherwise require a heavy expenditure to investigate. The drilling may also lead, and sometimes does, to the development of productive areas which, were it not for this relatively cheap method of exploration, might lie dormant for years. Favorable structures and any ore indications are followed up by excavational work.

METHODS OF SAMPLING

The underground sampling done may be divided into two broad classes, depending upon the purpose of each, namely (1) sampling used as a guide for current operations, and (2) sampling on which estimates of ore reserves are based. The former constitutes the bulk of the sampling and will be discussed first.

Development and Exploration Samples

Diamond Drill.- Sludge samples are taken and assays made for each 10 feet of hole. These samples are usually not of much value, but their cost is small in comparison with the cost of the drilling itself, and the occasional additional information afforded by them warrants the trouble and expense involved.

Diamond-drill cores are carefully examined and logged and are split for assay where at all promising in appearance. A single sample never represents more than 5 feet of core and, if changes in the material traversed are apparent, samples may represent as little as 6 inches of core.

Development Headings.- Crosscuts are rib sampled where any break or ore indication is traversed.

Daily face samples are taken from all drift faces as they progress. Any development rock encountered which will assay $2 or more per ton is sent to the mill, as that figure will cover the milling charge and the tailings loss, and will leave a small profit on the milling alone.

As drifting progresses, the preliminary sampling is followed up by channel sampling of the backs at 5-foot intervals to delimit any commercial ore shoots, as well as to determine any favorable locations for exploratory raises.

Stopes.- As stoping progresses, careful sampling of the walls is done when mining to an "assay wall." When the material is obviously of commercial grade enough breast samples are taken to serve as a rough check against the grade assigned to the ore block based on the back sampling of the drift. A close watch is kept on the walls to locate any branch veins leading off from the main body.

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Broken Ore to Mill.- As the ore is loaded at each working place (stope or drift), a grab sample consisting of a handfull of fines is taken from each car and deposited in a box, and at the end of the shift is taken up for assay. While the method of taking these muck samples is very crude and day-to-day samples are subject to considerable variation, the results obtained are of much value in regulating the mill feed and in indicating the grade of ore being currently obtained from various working places.

The value of the ore milled in the past year, as calculated from the muck sample assays, was 4.8 per cent below the actual mill recovery plus the tailings loss.

Estimation of Tonnage and Value

The cross-sectional value of an ore lens at any level is determined by averaging the back samples (channel samples cut from the back and at right angles to the vein) taken after the drift has been slashed out to the full width of the orebody, the samples being weighted according to their spacing and to the width of ore represented by each sample. This average value is calculated after obviously high samples have been averaged with the samples on either side and the resulting value substituted in making the calculation. In arriving at the tonnage and value of the block, the area and value of ore exposed at the level is assumed to extend 50 feet above and below the level, or is joined up with a corresponding exposure at the adjacent level if such an exposure has been developed.

This method of arriving at an estimate of ore reserves without the usual raises from level to level is adopted not because of the regularity of the deposits but rather because of the fact that the deposits are extremely irregular in cross section and in value. While the method often does not result in an accurate estimate of the ore contained in any given block, experience has shown that losses in one quarter are offset by gains in another, so that the method at any given time results in a reasonably accurate picture of the mine as a whole.

Ore reserves are calculated annually, at which time any new blocks are included on the basis outlined above. Partly mined blocks are included for the calculated tonnage remaining unmined and at the value originally assigned to the block, unless the mining accomplished has shown this figure to be obviously incorrect, in which case a new calculation for grade is made. Broken ore is included at a value 10 per cent below the calculated value of the ore in place, unless the "box samples" (grab muck samples) taken in the course of extraction have shown this figure to be obviously incorreot, in which case an adjustment is made, giving due consideration to the value indicated by the muck samples.

METHODS OF DEVELOPMENT AND MINING

General

The property has been developed and is now being worked through a main vertical shaft sunk to a depth of 1,200 feet. Working levels are established at the 100, 200, 300, 400, 500, 600, 733, 866, 1000 and 1200 foot horizons. The shaft is 6 feet by 15 feet 2 inches outside the timbers. It is timbered with 8 by 8 inch square timbers and is divided into three compartments, each being 4 feet 8 inches by 4 feet 2 inches in the clear. Two compartments are used for hoisting (cages in counterblance) and the third compartment is used for a ladderway, pipe lines, and electric cables. The present active mine workings are

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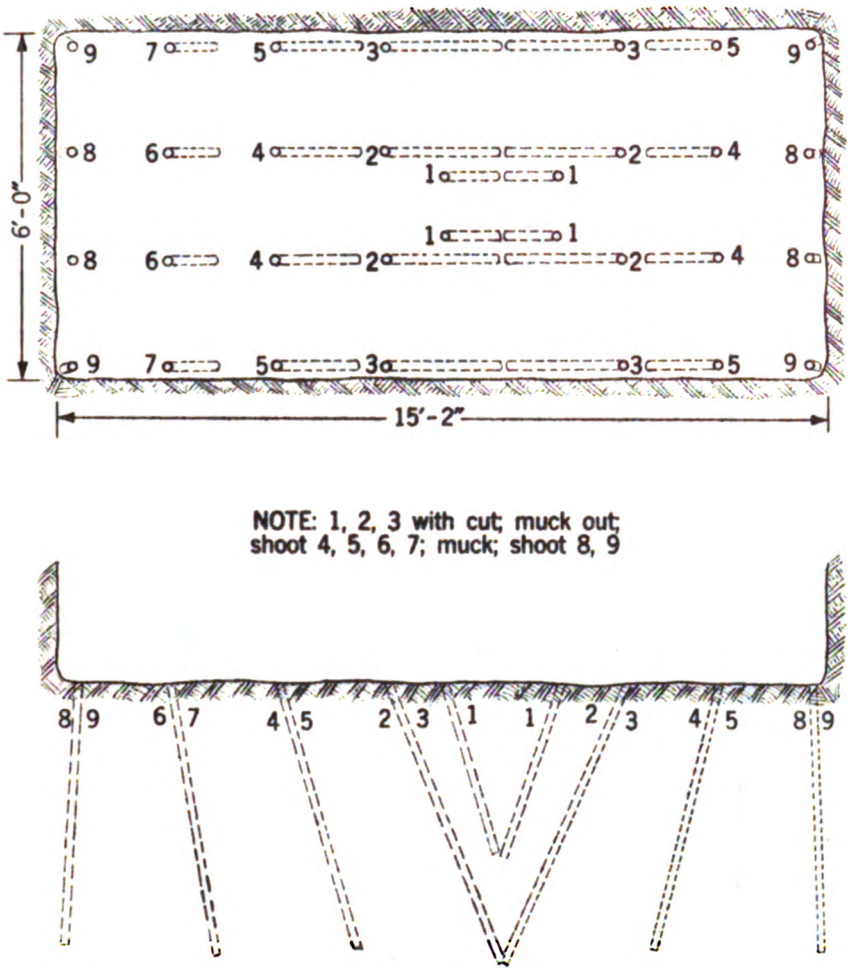

Figure 3.—Standard shaft round

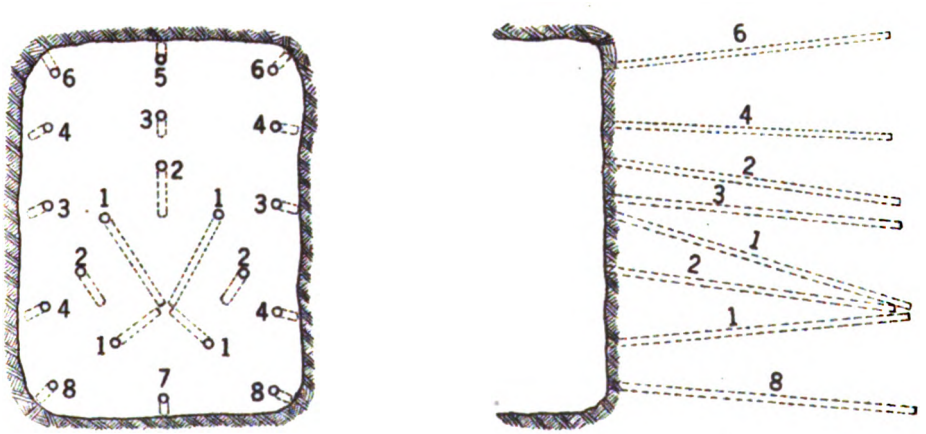

Figure 4.—Standard drift round

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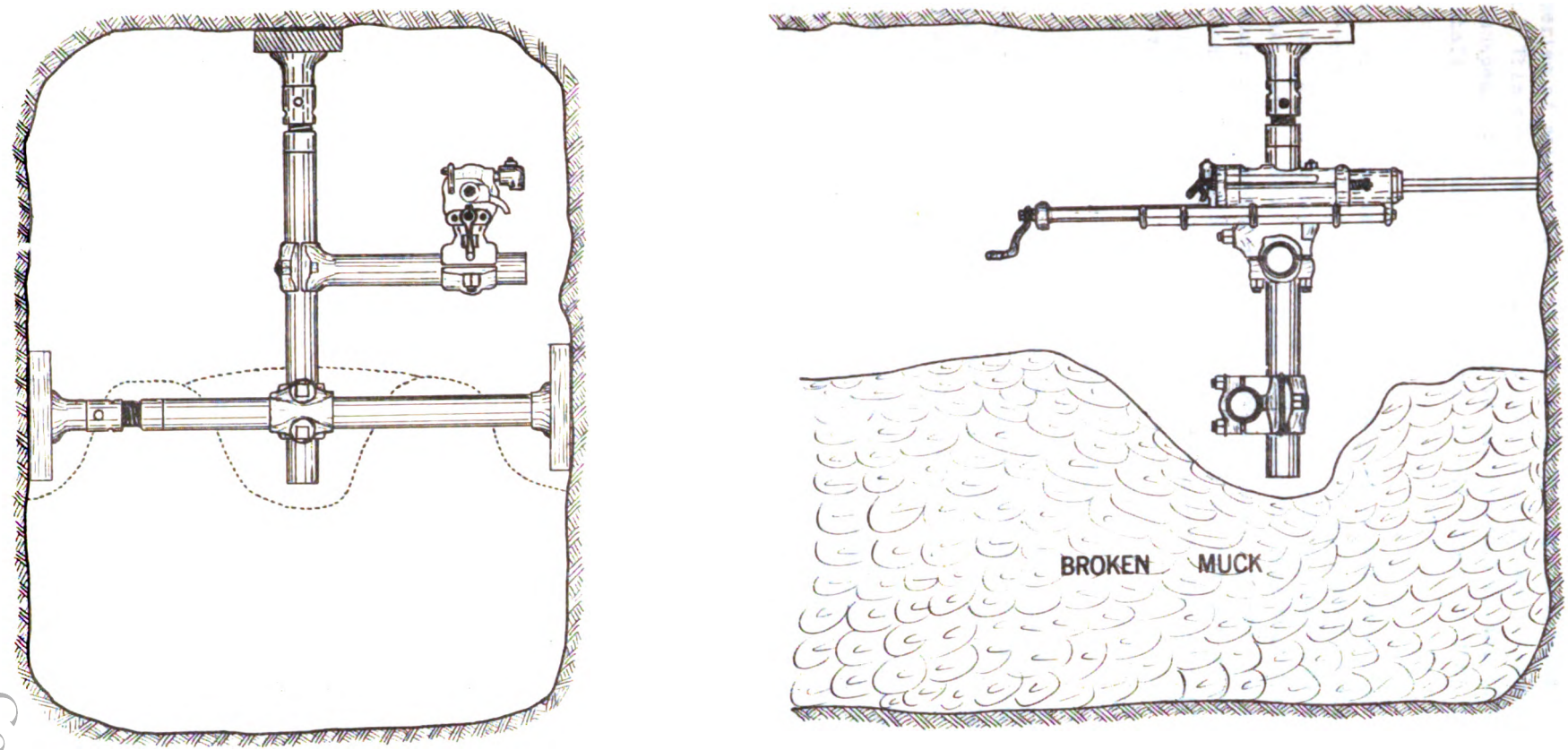

Figure 5.—Crossbar and short column drill mounting

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connected by a series of raises and crosscuts to the Old Vipond workings, the shaft of which is 400 feet deep. This shaft serves as an auxiliary exit in case of emergency, as a separate route entirely independent of the main shaft is maintained from the 1200-foot level to the surface. The shaft is of additional benefit at all times in that it greatly improves ventilation.

Current work in the development of the mine is all done from the North Thompson (main) shaft. From each level of this shaft, crosscuts are driven to the breaks or veins in which ore lenses are known to occur, and drifting is done to determine the payable portions which are later extracted. From the drifts and crosscuts resulting from this work, a diligent search is made by diamond drilling and by subsidiary drives for new lenses not previously exposed on other levels. The irregularity of the ore occurrences as shown by Figures 1 and 2, previously referred to, makes it apparent that no uniform lay out for all levels is possible.

Development Details

Shaft.- The shaft is at present being deepened below the 1200-foot level. A standard shaft round is shown in Figure 3, the numbers indicating the order of firing. Owing to limitations in hoisting capacity, it is necessary to do the work in 24-hour cycles, which must be strictly adhered to. A cycle is as follows:

The day shift (7:00 a.m. to 3:00 p.m.) drills off the round and blasts the cut, holes No. 1, 2, 3 and removes the resulting muck. They reblast the cut if necessary and blast holes 4, 5, 6, and 7 when coming off shift. The afternoon shift (3:00 p.m. to 11:00 p.m.) mucks as much as possible and when coming off shift blasts the remaining holes of the square up. The graveyard shift (11:00 p.m. to 7:00 a.m.) clears the shaft of muck, picks the bottom, lets down the machines and hose, and makes the air and water connections ready for the drilling to be done by the day shift. All the drilling is done on the day shift and by the same crew of five men operating drifters with 3½-inch cylinders, equipped with handles and used as pluggers. The crews, composed of five men each on the afternoon and the graveyard shifts, change shifts each week. All the men employed in the shaft are miners and do their own timbering. A 5-foot round is pulled six days per week and the timbering is done on Sunday. Blasting is done by electric time fuse and battery.

Drifts and Crosscuts.- A standard drift and/or crosscut round is illustrated in Figure 4. Cut holes No. 1 are blasted first and if it is found necessary they are reloaded and blasted with the square up. Where possible, a clear heading is furnished for drilling: that is, an advance of one round per 24 hours is made.

In certain development headings, which it is desired to rush, an advance of two rounds per 24 hours is made. Figure 5 shows an arrangement of crossbar and short column which has been found useful in avoiding "mucking back" to clear the heading for drilling, where advancing two rounds per 24 hours. The crossbar is set in place about 3 feet below the roof of the crosscut and above the muck from the previous round. To the crossbar is fastened, by means of a universal clamp, a short column which is jacked to the roof. Drilling is done off the vertical column in the usual way, the machine itself being mounted on a standard arm clamped to the short column. By the time the top half of the round is drilled off, most of the muck has been removed, and the column is turned on the crossbar and jacked to the floor, and the drilling completed. This arrangement, which may seem a bit cumbersome,

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is gladly used by the miners in preference to mucking back for the usual set-up of a column, or making two set-ups of a crossbar. It has undoubtedly added to the ease and certainty of regularly removing the two rounds daily from our double-shift headings.

The usual rate of progress is 5 feet per day for single-shift headings and 10 feet per day for double-shift headings. All mucking is done by hand. A crew consists of a runner, a helper, and two muckers. Blasting is done by cap and fuse.

Raises.- For a raise of 6 by 6 feet, a round similar to the standard drift round is used. For a smaller raise, and especially if in quartz, a "burned cut," illustrated in Figure 6, is used. All raises are driven on an angle of 50 to 55° and without any timber. Usually they are driven from the backs of stopes for a distance of 30 to 50 feet to connect with the level above. When driven from level to level on the vein for exploration purposes, they are driven at the same inclination and without timber, except for the chute and manway at the bottom. This arrangement is illustrated in Figure 7.

Drill Steel Used

One-inch quarter-octagon drill steel with a ¼-inch round hole is used. The gage of the bit for various lengths is as follows:

	Length, feet	Gage of bit, inches
Starter	3	1⅝
Second	5	1½
Third	7	1⅜
Fourth	9	1¼

Figure 8 shows the bit used, which is a single taper cross bit, and also shows the shank used.

It will be noted that the hole in the steel is punched out to a 9/16-inch diameter at the cutting edge of the bit. This feature improves the drilling speed of the bit somewhat, as it tends to the production of coarse cuttings. It also aids materially in maintaining the gage of the bit, as it keeps the center of the hole high, which tends to keep the drill centered in the hole without excessive wear on the outside cuttings edges of the bit.

The drill is shanked in the sharpener to the form shown in Figure 8 by use of an old dolly machined out to the proper shape. The diameter of the surface which takes the blow of the hammer is made 15/16 inch (1/16 inch less than the smallest diameter of the steel), so that a slight upsetting of the shank will not cause the steel to bind in the chuck of the machine.

Mining

Stoping.- Shrinkage stoping is the method of stoping exclusively used at Vipond. As the ore is broken, casual pillars of ore are left where absolutely necessary, and any waste or nonpayable material encountered in the vein is left and serves to help support the walls while the stope is being scaled and the broken ore removed.

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NOTE: Shoot 1 until cut is out; sequence 1, 2, 3, 4, 5

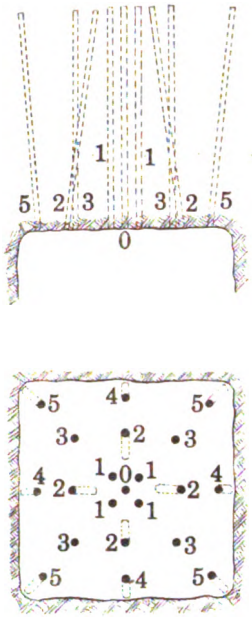

Figure 6.—Burned cut in raise

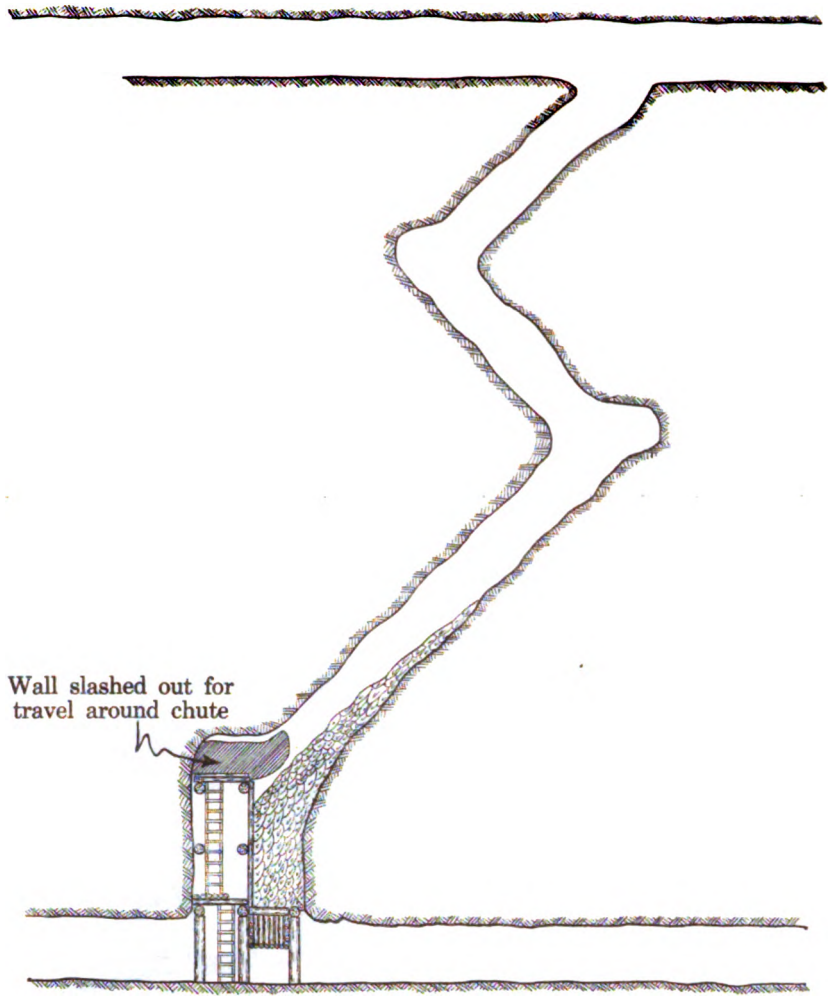

Figure 7.—Exploration raise and chute

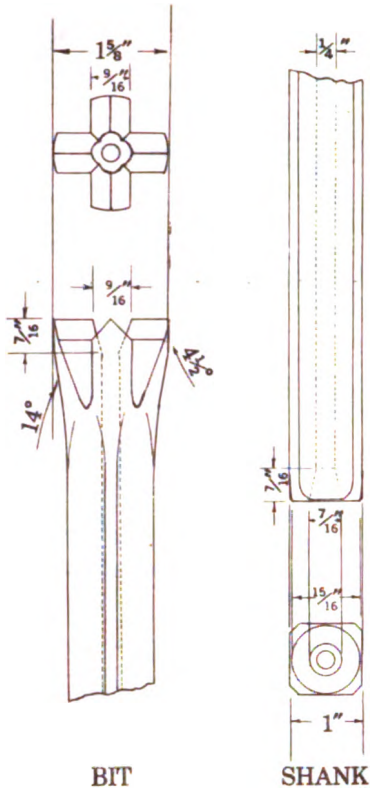

Figure 8.—Drill bit and shank

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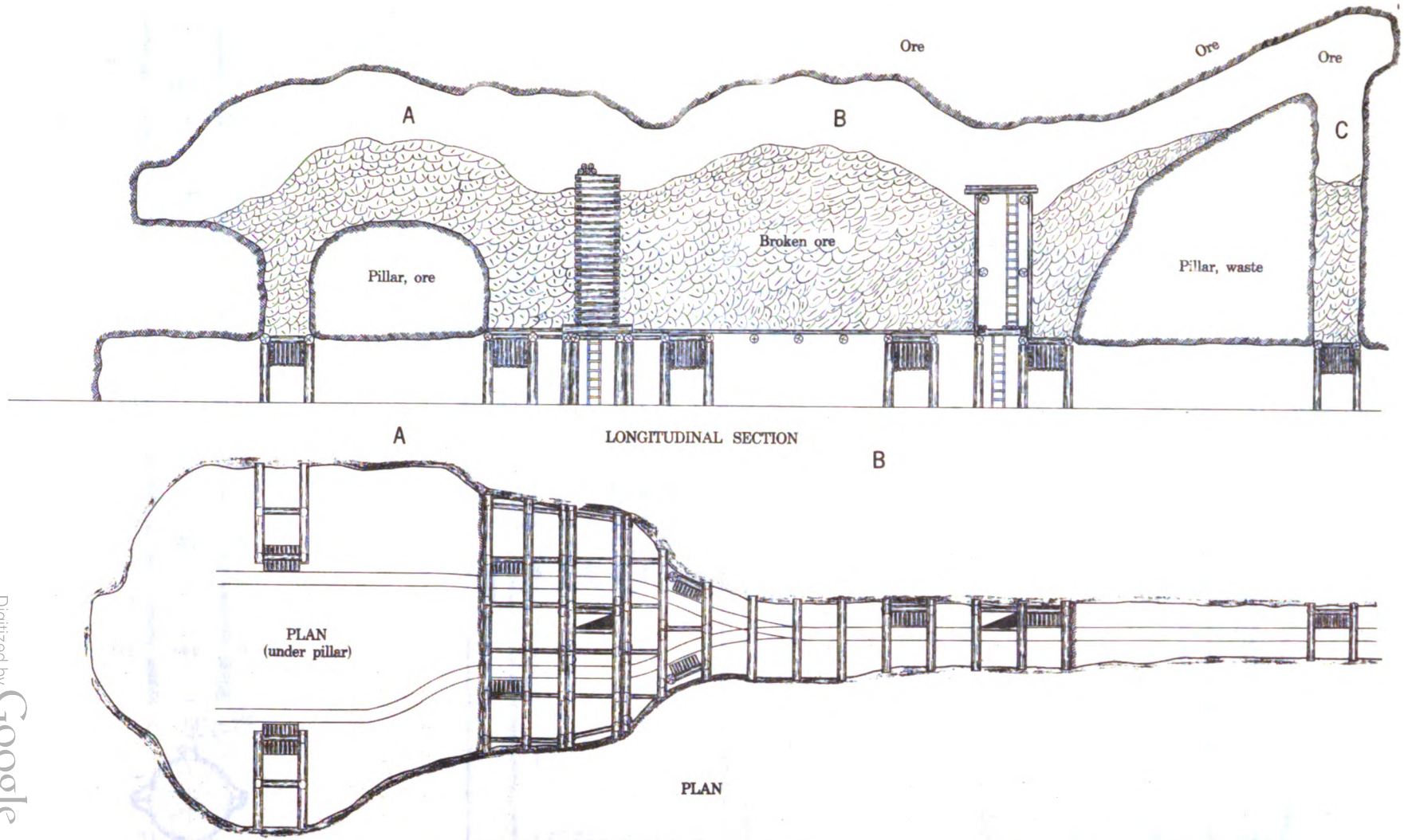

Figure 9.—Stoping in an ore lens of varying width. A, Stoping with box-hole pillars; B, stoping on timber; C, prospect raise up into ore

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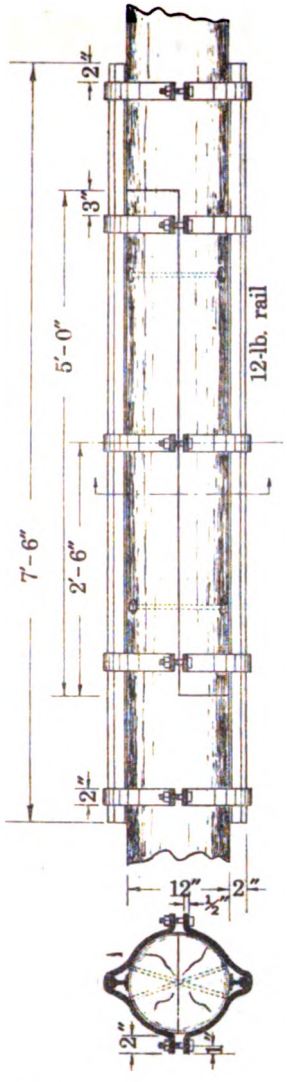
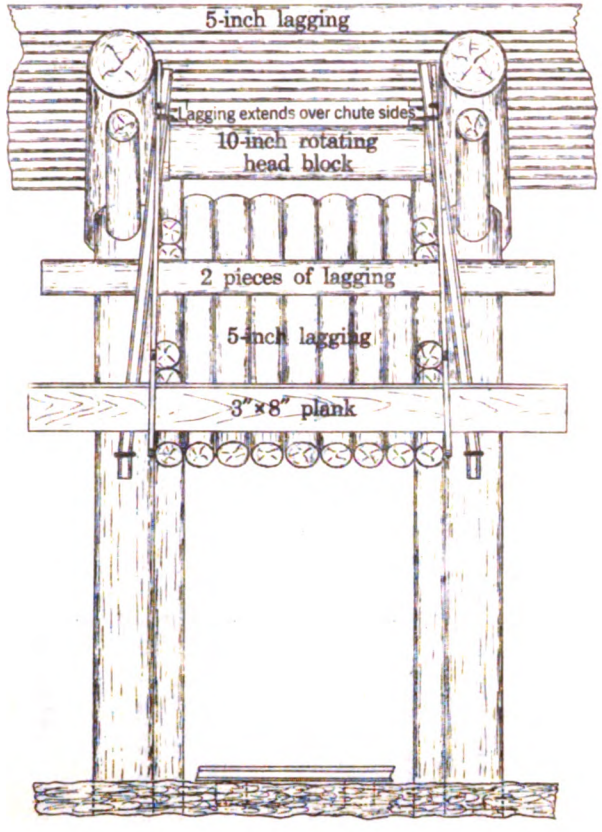


Figure 10.—Method of splicing stull timbers



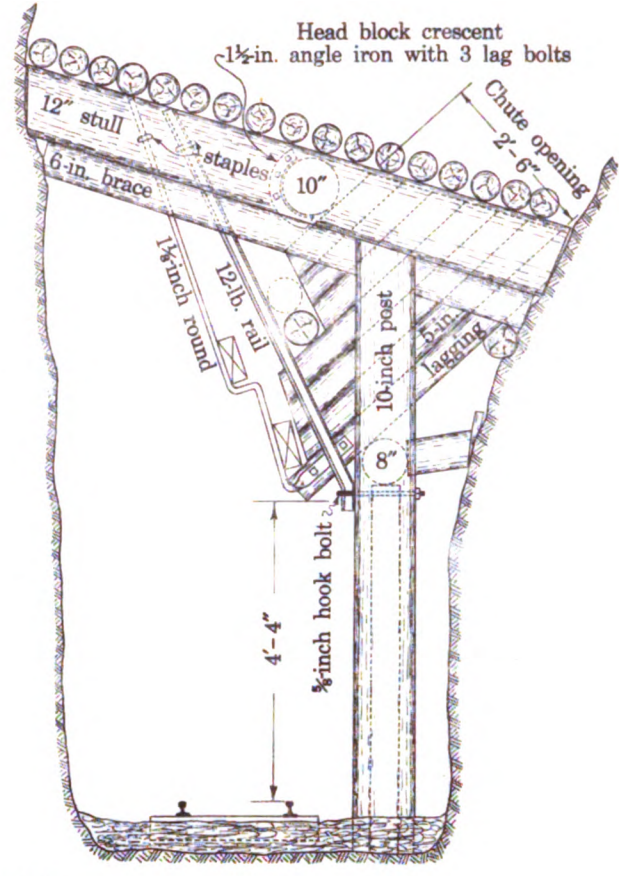


Figure 11.—Standard ore chute

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The choice between supporting the broken ore on stulls and lagging or mining it through box-hole raises depends upon the width and grade of the ore. Narrow veins if of payable material are always stulled and lagged, the stulls being supported in hitches. If the vein material exposed at the level is not payable but is of low grade, exploratory raises are driven at favorable points; and if these raises encounter payable ore, a stope is started and the resulting ore drawn off through these raises and any others that it might be necessary to drive for extraction purposes. Ore-bodies up to 25 feet wide are stulled and lagged, the stulls being supported wholly by posts. Ore-bodies in excess of 25 feet in width are opened up by box-hole raises and the roof and floor pillars extracted after the stope is drawn empty.

Figure 9 shows an ore lens of varying width in which each of the methods just described are used.

Where timbering is used, the chutes are placed at 15-foot intervals. Where box holes are put up, they are rarely placed closer than at 25-foot centers, and if they must be driven far to encounter the ore, they may be 40 feet or even more apart.

Each stope is served by cribbed manways at 100-foot intervals. The stope is carried up about 60 feet, after which a raise is put through to the level above and all but one of the manways covered over. The breaking of ore then proceeds through to the level above.

As the broken ore is drawn from the stope, the walls are scaled and bad rock is supported where necessary by sprags or stulls. In narrow stopes only small short sprags are required. In the wider portions more scaling is done to avoid the use of large quantities of heavy timber. Where an occasional exceptionally long stick is required, a splice of two sticks is made as shown in Figure 10 though this expedient is not resorted to if it can be avoided. A stick spliced as shown has very little strength as a beam, but has been successfully used as a sprag, up to 35 feet in length.

The relatively good walls has made it possible to scale all stopes and muck them clear of broken ore. This procedure involves considerable labor but results in the complete removal of the broken ore and in an additional yield from stringers and small masses of ore which were left in the walls as mining progressed.

Mining in stopes is done with Leyner machines drilling horizontal holes. Scalers are equipped with pluggers.

UNDERGROUND TRANSPORTATION

All loading and tramming is done by hand labor. Figure 11 shows the type of chute employed. Ore is transported in flat-bottomed, end-dump cars fitted with Timken bearings. The cars, having a capacity of 1 ton each, are hoisted to the surface on cages operating in balance. The ore is dumped directly into the primary crusher which is located at the shaft head, and the waste is trammed to the waste dump.

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PER CENT EXTRACTION

The extraction of the ore is complete except for any pillar or pillars which may be left in the course of mining and which can not later be recovered. Most stopes are finished without leaving a single pillar. The recovery will exceed 98 per cent.

No waste is sorted from the ore either below ground or on surface. The dilution of the ore by waste amounts to about 10 per cent.

COMPARISON OF DIFFERENT METHODS USED IN THE DISTRICT

Until comparatively recently, shrinkage stoping was practically the universal practice in the district. As mining reached greater depths, it was found that the walls did not stand well, and attention was given to other methods of mining. A few stopes were taken out by square-setting but now the cut-and-fill method is becoming the more general practice in those cases where shrinkage stoping is not used.

WAGES PAID, CONTRACT AND BONUS SYSTEMS EMPLOYED

All work below ground except in drifts and crosscuts is done on days pay. The wage rate is: Shift bosses, $7 per shift; miners, pumpmen, scalers, trackmen, cage tenders and powdermen, $4.80 per shift; timbermen, $4.80 and $5.05 per shift; muckers, helpers for machinemen, and helpers for timbermen, $4.24 per shift.

The driving of drifts and crosscuts is done on a bonus system. At one time the work was done on a so-called contract basis by which workmen were guaranteed the regular scale of wages but were given the work to do at a price per foot; they furnished the explosives and the labor for drilling, mucking and tramming. Unless the price paid was frequently changed to meet the varying conditions encountered, it was found that circumstances over which the workman had no control either made it impossible for him to make any bonus at all, or the bonus worked out at a figure which was obviously more than a just reward for any special skill and effort which he might bring to the task. The company found itself in a position where it suffered all the extra cost due to any adverse conditions encountered and did not benefit by unusually favorable conditions, if and when these were encountered. The system now employed does not entirely eliminate this difficulty, but it does to some extent, and it does reward good work under all conditions and at the same time leaves the company in a position to share in the benefits when the work is being done in easy ground.

The bonus now paid is for footage advanced in any month in excess of 4 feet per shift worked, and is as follows:

The drill crew is paid $4 per foot for all footage in excess of an average of 4 feet per shift worked. An allowance of $2.50 per foot advanced is made for explosives and the crew is credited with one-half of any saving they make, or charged with one-half of any explosives used in excess of this amount. The bonus due the drill crew on the above basis is then split three-quarters to the drill runner and one quarter to the helper.

The muckers are paid 50 cents each for footage in excess of an average of 4 feet per shift.

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VENTILATION

Ventilation of the mine is almost wholly natural, with the aid, of course, of the compressed air, which is exhausted from the machines while running, and the usual "blowing of smoke" by compressed air between shifts after blasting. In addition to this, a 10-inch fan operated by electric motor has been used on occasion on the lower levels to increase the circulation of air from the shaft through raises or stopes to the upper levels.

The North Thompson or main shaft is downcast, whereas the Vipond shaft, which is at a somewhat higher elevation, is upcast. In the summer months, little or no attention need be given to the rate of air circulation, but in the winter months the natural circulation of air must be checked by means of ventilation doors to avoid trouble with ice in the downcast shaft. It is not unusual for ice to form down to the 500-foot level of the main shaft, and care must be taken to avoid trouble with the cages.

FIRE HAZARDS

The fire hazard in the mine is not considered to be great, though it is recognized that such a hazard does exist in spite of all the care which may be taken to minimize it. The shaft is timbered but is always wet. The stopes require little timber and the ore itself is not inflammable.

To cut down the risk of fire, only one day's supply of explosives is taken below ground each day, all oil is taken below and kept in metal containers, electric wiring is carefully done and frequently inspected, and no inflammable refuse is allowed to accumulate below ground. Cleanliness of workings is insisted upon and made effective as far as possible

Equipment for fighting fire in the headframe, which is of wood construction, consists of fire extinguishers and fire hose connected to the mine hydrants. Little or no provision is made for fighting fire in the mine, though provision has been made to localize the effects of a fire should one occur, and to warn the workmen to leave the mine in case of danger.

At each level provision is made to close the level off from the shaft by means of a fire door. The door is light enough to be closed by one man and hangs from the roof of the drift so that it is out of the way, and there is little danger of the door being abused or damaged so that it can not be used if and when wanted. Various squads of miners are given training in mine rescue work, and the use of standard apparatus at the Provincial Mine Rescue Station located at Timmins.

Provision is made for the introduction of the stench ethyl mercaptan into the compressed air lines to serve as a warning to underground workmen to vacate the mine in case of danger. The United States Bureau of Mines gives an account[5] of the introduction of various stenches into air lines to serve as a danger signal. A consideration of their findings led to the development of the apparatus shown in Figure 12. These bomb breakers, each charged with a stench bomb, are installed on the main air lines leading to the shaft, and are always ready for instant use. This form of apparatus has the advantages of being cheaply and easily made at any mine, of being absolutely effective as to the introduction of the

5 Katz, S. H., Allison, V. C., and Egy, W. L., Uses of Stenches as a Warning in Mines: Tech. Paper 244, Bureau of Mines, 1920, 31 pp.

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stench, and of being so constructed that it may be relied upon to work without frequent inspection.

A test of the apparatus and the stench ethyl mercaptan was made under working conditions at the mine and the results observed by officials of the Inspection Branch of the Ontario Department of Mines. The Ontario Department of Mines Bulletin 71 gives a full account of the results and observations. The test showed the warning signal to be effective.

SAFETY METHODS AND FIRST-AID ORGANIZATION AND TRAINING

In the operation of the mine, the code of rules contained in the Mining Act of Ontario, 1930 (part 8, and secs. 178, 179, and 180 of part 9), are adhered to. No regular inspection is provided other than that done by the mine captain and the shift bosses. Groups of employees are from time to time trained in the practice of first aid, under the auspices of the St. John Ambulance Association. A first-aid kit is kept at the office. As the mine is located near the town, doctors and hospital accomodation are readily and promptly obtainable.

Table 1.- Summary of Costs

Name of mine: Vipond mine. Period: August 1, 1929 to July 31, 1930.

Tons of ore hoisted during period: 113,329. Tons broken: 104,381. Mining method: Shrinkage.

Underground costs per ton of ore hoisted

	Labor	Supervision	Compressed-air drills and steel	Power cost	Explosives	Timber	Other supplies	Total
Development	$[1]0.362	0.044	0.128	-	0.151	-	0.043	0.728
Mining (stoping)	.626	.084	.250	-	.180	0.046	.089	1.275
Transportation (underground)	.545	-	-	0.046	-	-	.038	.629
General underground expense	.042	-	-	.045	-	-	.061	.148
Surface expense (directly applicable to underground operation)[2]	-	-	-	-	-	-	-	-
Total	1.575	0.128	0.378	0.091	0.331	0.046	0.231	2.780

1 Includes diamond drilling at 8.1 cents per ton.
2 Distributed in various underground accounts.

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Table 2.- Summary of Costs in Units of Labor, Power and Supplies

Name of mine: Vipond mine. Period: Aug. 1, 1929 to July 31, 1930.

Tons ore mined and hoisted: 113,329. Tons broken: 104,381.

	Development	Mining (stoping)	Total
A. Labor (man-hours per ton)			
(Breaking (drilling and blasting)	0.212	1.176	1.388
(Timbering			
Shoveling	[1] .212		.212
Haulage and hoisting			.970
Supervision			.107
General			.149
Total labor underground			2.826
Av. tons per man per shift			2.830
Labor, percentage of total cost			56.66 per cent
B. Power and Supplies:			
Explosives (lbs. per ton)	0.73	1.05	1.78
(kind and grade)	(97% - 40 per cent gelatin (3% - 55 per cent ammonia		
Timber (b.m. - lin. ft. or cu. ft.	(.42 bd. ft. (.52 lin. ft. lagging (.18 lin. ft. stulls		
Total Power (kw.h. per ton)			28.0
(1) Air compression	7.1	11.8	18.9
(2) Hoisting	1.1	3.5	4.6
(3) Pumping			3.9
(4) Lighting and Miscellaneous			.6
Other supplies in percentage of total supplies and power			46.0 per cent
Supplies and power, percentage of total cost			43.34 per cent

1 Includes tramming development muck.

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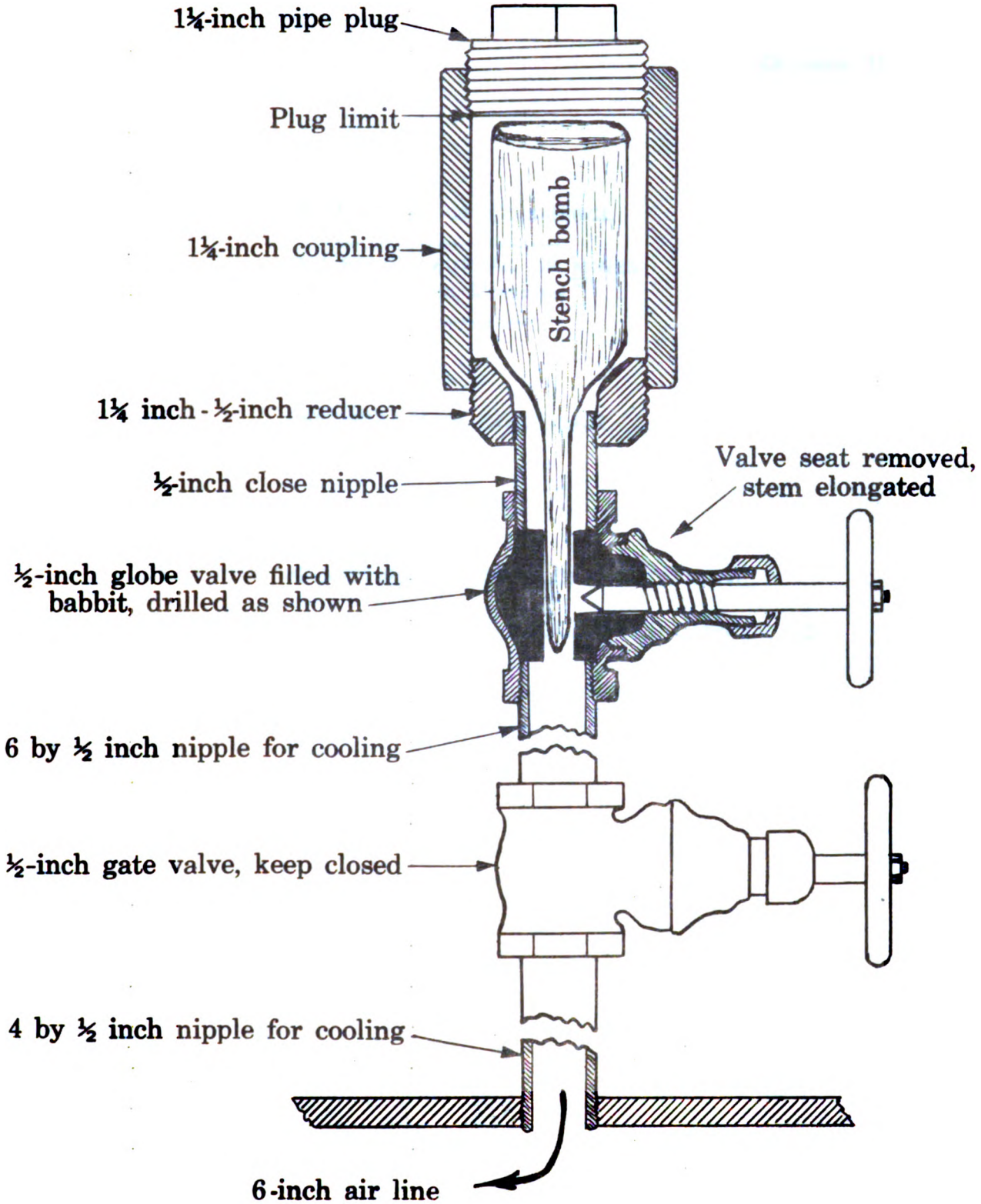
1¼-inch pipe plug
Plug limit
1¼-inch coupling
Stench bomb
1¼ inch - ½-inch reducer
½-inch close nipple
Valve seat removed, stem elongated
½-inch globe valve filled with babbit, drilled as shown
6 by ½ inch nipple for cooling
½-inch gate valve, keep closed
4 by ½ inch nipple for cooling
6-inch air line

Figure 12.—Stench bomb apparatus

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